METALLURGICAL TREATMENT AT MOUNT MORGAN, QUEENSLAND.

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INTRODUCTION.

The ores treated at Mount Morgan are divided into three classes, viz.: (1) oxidised ore, (2) mundic ore, (3) copper-gold ore.

The following represent partial analyses of these ores:—

—	Oxidised Ore.	Mundic Ore.	Copper-Gold Ore.
SiO_2	88 %	63-75 %	45-49 %
Fe	4·5%	15 %	23 %
S	·5%	15 %	22 %
Cu	Traces	·4-1·7%	2·8-3·7%
Au	3 dwts. and over per ton.	8-15 dwts. per ton.	7-10 dwts. per ton.

The oxidised ore is a highly silicious ore, the mundic ore is a silicious sulphide ore with little copper, and the copper-gold ore is a sulphide ore, with less silica than the previous ore, and containing both gold and copper. The change between the oxidised ore and the mundic ore takes place at varying levels in the open cut. At lower levels the mundic ore gives places to the copper-gold ore. Each class of ore is treated separately in three different works, viz.: The West Works, the Mundic Works, and the Copper Reduction Works. At the time of my visit,—May, 1910—the West Works were just about to shut down, all payable oxidised ore having been worked out.

The oxidised ore is obtained from the open cut by a system of benching and milling. The mundic ore is partly obtained from the open cut, but mostly from underground by a system of stoping, using square sets and filling. The copper-gold ore is obtained from underground by a similar system.*

* Transactions Australian Institute of Mining Engineers, 1910.—"Underground Mining Method at Mt. Morgan," by White, Sewell & White. "On Recording and Estimating," by Patterson Thomas.

Constant assaying is necessary to decide whether the ores are of sufficient value to be treated, and whether the sulphide ores are to be classed as mundic, copper-gold, or poor.

METHOD OF SAMPLING.

Underground.—Samples are taken from the drill-holes per each set in stoping, and per each cut in driving and sinking, and sent to the assay office. Samples required to be assayed as soon as possible, are marked with a special label, and they are given precedence over all others.

Open Cut.—A box is placed near each spot where the men are filling trucks, either from the bench floors or from passes, and into this the men throw a small handful of ore off the top of each truck as they fill it. This sample is then divided and quartered down to a few pounds, and then sent to the assay office. All the mine samples are assayed for copper, silica, and gold.

SMELTING WORKS.

As the trucks are discharged into the smelter bins, 1-23 of the ore, etc., is removed by means of a narrow shoot, 4in. wide, running crosswise under the trucks. This shoot discharges its contents on the floor under the bins. Three men, each with a charge barrow, load at the same time, each man throwing a shovel-ful into each truck in rotation. One truck goes to the sampling mill, where it is crushed and sampled to small bulk, the other two to the furnaces.

AVERAGE OF ALL ASSAYS FROM 1907 TO 1910, FOR COPPER-GOLD SULPHIDE ORE.

——	Gold.	Copper.	Silicia.
Stope Assays	7·98 dwts.	3·53 %	43·1%
Smelter Assays	8·00 ,,	3·49	45·9
Difference	·02 ,,	·04	2·8

All the payable ores are trucked to the main shaft, there hauled, tipped, and conveyed to separate large hoppers. From these large hoppers they are conveyed in trucks hauled by electric motors to the main storage and reciving hoppers, or bins, at the different works.*

* Transactions Australasian Institute of Mining Engineers, 1910.—"Head Gear and Winding Plant, etc." by N. F. White.

Treatment of the Oxidised Ore at the West Works.—Capacity of works, 400 tons per day. The receiving hoppers at the works have an aggregate capacity of 2,000 tons. From these hoppers the ore passes along a shoot into the breakers placed immediately below. The plant was designed and built in four sections, so as to commence the treatment of the ore as soon as possible.

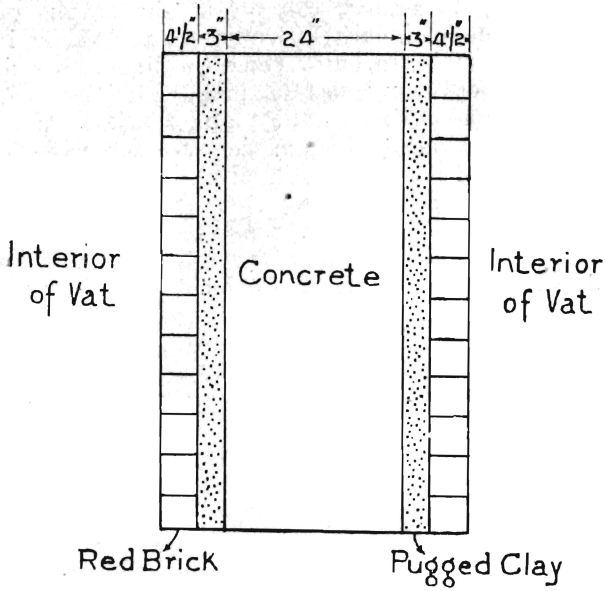
No. 1 section has a 24in. by 16in. Blake-Marsden jaw breaker, the remaining three sections, rotary machines of the Jaques type (No. 4), making 420 revolutions per minute. Both types have given equal satisfaction. The ore is reduced in these breakers to from 1in. to 2in. fragments. It then falls directly into inclined shoots into rotary cylindrical dryers, of which there are four, one to each section. The ore, as it comes from the mine, runs up to 12 per cent. moisture, and it is necessary to drive most of this off, otherwise the efficiency of the ball mills would be greatly impaired. Even chemically combined water has the effect of making finely-crushed ore choke the screens. The cylindrical dryers are 30ft. 5in. long, 6ft. 8in. diameter, and they are set at an inclination of $\frac{3}{4}$in. to the foot. They are made of mild steel plates, $\frac{1}{2}$in. thick, and are lined throughout with fireclay bricks made on the company's works. The internal diameter of the dryer is 5ft. 4in. The shell is built in three sections, united by cast-iron tyres rivetted to the shell. These tyres are flanged and each rests on a pair of friction rolls, by means of which the dryer is made to revolve. The rolls are 30in. in diameter, and have 8$\frac{1}{2}$in. faces.

The dryers revolve at about one revolution per minute. The fire-box is at the lower or discharge end, and is fired either by wood or coal. The intensity of the fire is regulated by the amount of ore supplied and the moisture it contains. When the ore is dry, no fire is put on, and the dryers then act simply as conveyers. The dried ore leaving the dryers falls into a chamber at the end, and is there caught and elevated by means of four chain and bucket elevators, moving about 5ft. per second, and is discharged into storage bins above the ball mills. From these bins the ore passes through shoots to sixteen No. 5 Krupp ball mills, crushing through brass wire gauze screens to 400-600 mesh. These mills are hand-fed, and the average mill revolutions per minute are twenty-two. The balls are 6in. in diameter, and their average total weight in each mill is about one ton. The balls and grinding plates wear away at the rate of 2$\frac{1}{4}$ pounds per ton of ore crushed. From the ball mills the crushed ore passes to the revolving furnaces storage hoppers, from which it is fed mechanically by self-feed scoops into four revolving roasting furnaces. Each furnace is 43ft. long by 7ft. 6in. in diameter for the first 16ft. at the lower end, and 6ft. 8in. for the remaining length. Each is built of $\frac{1}{2}$in. mild steel plates, lined with fire-bricks, so that the internal diameter of the furnace is 5ft. 4in. These furnaces are set at an inclination of $\frac{1}{2}$in. to the foot towards the discharge end. They rest on four pairs of friction rolls as is the case with the dryers, and are driven by pinion and spur gearing, making one revolution in 100 seconds.

The roasted ore falls into passes on either side of each furnace, and is picked up by a revolving combination cooler and conveyors, one on each side of each furnace. These coolers and conveyors are made of $\frac{3}{8}$in. steel plates in three sections, with two cast-iron tyres. They revolve on two pairs of rolls, 14in. diameter, and are worked off the same shaft as the furnace, being driven by bevelled gearing at a speed of two revolutions per minute. They are fitted internally with worms of 16-inch pitch. Bolted to the outside of the cylinder are five slotted pipes, 9ft. long and 2in. in diameter. These pipes take up water from a tray underneath, and discharge the same over the heated surface of the conveyor at each revolution. The furnace is fired with ironbark billet wood, the amount used being about $2\frac{1}{2}$ cwts. per ton of ore roasted. In addition to wood firing, the furnaces are fired by producer gas. This gas is produced in two water-sealed gas producers, each 11ft. 10in. in height and 13ft. in diameter, built of $\frac{3}{8}$in. plate steel and brick lined. The amount of coal used in the producers is 12 cwts. every three hours, and averages 1.4 cwts. per ton of ore roasted.

The roasted ore is taken by the conveyors and stored in brick hoppers, from which it is taken to the open vats by means of a four-ton ending tipping truck, running along the vat walls, at a cost of about 3d. per ton. There are 16 vats in all, 4 to each section; each vat measures 60ft. by 12ft. by 5ft. deep. They are built of concrete. The four vats in each section are arranged in a group, two side by side, with a wall dividing them for their whole length, and two end on to these. The floor of the vats is made of concrete 18in. thick, and has a fall of 3in. from end to end. The walls are made of concrete 2ft. thick, then 3in. of well pugged clay and then $4\frac{1}{2}$in. of brickwork as per figure.

This clay keeps continually moist, and effectively protects the concrete work from the varying temperatures of the newly charged ore, and also from the chemical action of the liquors. The filter-bed is made up thus—bricks on narrow side, then bricks on flat across these forming a grid, then 6in. of gravel of diminishing dimensions and sand, then a layer of perforated hardwood boards. The total height of the filter-bed is about 15in. to 18in. above the bottom of the vat. Each vat has a capacity of 120 tons.



Section through dividing Wall

As soon as the vats are filled with hot, roasted ore, they are filled with water applied at the bottom and also a little sprayed on with a hose at the top, the charge being thus finally cooled prior to the application of the chlorine solution, also soluble iron salts are leached out. After this water wash has drained off, the surface of the ore in the vat is raked, and chlorine water is run on from the main through a branch pipe with a length of rubber with a screw clip attached. This chlorine water contains about 80 grains of chlorine to the gallon, and it is run on until all the gold has been dissolved.

$$2Au + 2H_2O + 3Cl_2 = 2AuCl_3H_2O.$$

This point is ascertained by the tester in charge of the chlorination floor, taking a test tube three-quarters full of the liquor and adding to it sulphate of iron solution. This reagent is made up fresh every day by simply dissolving iron turnings in sulphuric acid and filtering through cotton wool.

$$2AuCl_3 + 6FeSO_4 = 2Au + Fe_2Cl_6 + 2Fe_2(SO_4)_3$$

When the vat liquors give only a slight reaction with this reagent, the wash waters are turned on until all soluble gold chloride has been washed out. The residues are sluiced out at a cost of one penny per ton with waste liquors at 50lbs. pressure per sq. in.

The consumption of chlorine per ton is about 1¾lbs., at a cost of 5½d. per lb.

The chlorine-gold liquors run off at the bottom of the leaching vats through diaphragm lead-antimony valves into a 9in. lead-antimony main, through which the liquors are drawn by three 9in. lead-antimony vacuum pumps, delivering into a sump well. The lead-antimony alloy used assays 95 per cent. Pb., per cent. Sb. From this sump they are forced into four 10,000 gallon circular cement-lined brick tanks, from which a regulated flow is admitted to the charcoal-packed filters for precipitation of the gold contents. As these liquors still contain free chlorine, steam is admitted into the tanks, which heats up the liquors and expels the chlorine. The warm solution then passes to the filters.

These filters, twelve in number, are rectangular, brick built, cement lined, and coated inside with a bituminous paint. Each one measures 9ft. 10in. by 10ft. 9in. by 4ft. 6in. deep, and holds about two tons of crushed charcoal. The filters are made up thus—perforated boards on the bottom, then a layer of cheese cloth, then charcoal, free from dust, 30 cwts. to 40 cwts., then perforated boards, and finally a layer of sand. The charcoal is first broken into fragments and sieved through a revolving screen into coarse, fine, and dust, the coarse charcoal through 3-16in. mesh the fine through 1-16in. mesh.

Seventy-two bags of charcoal yield about sixty-six bags of charcoal fit for use, the dust being thrown away. The charcoal is put into the filters in three layers—(1) 5in. of coarse on the cheese cloth, (2) then 2ft. of fine, (3) finally 5in. of coarse.

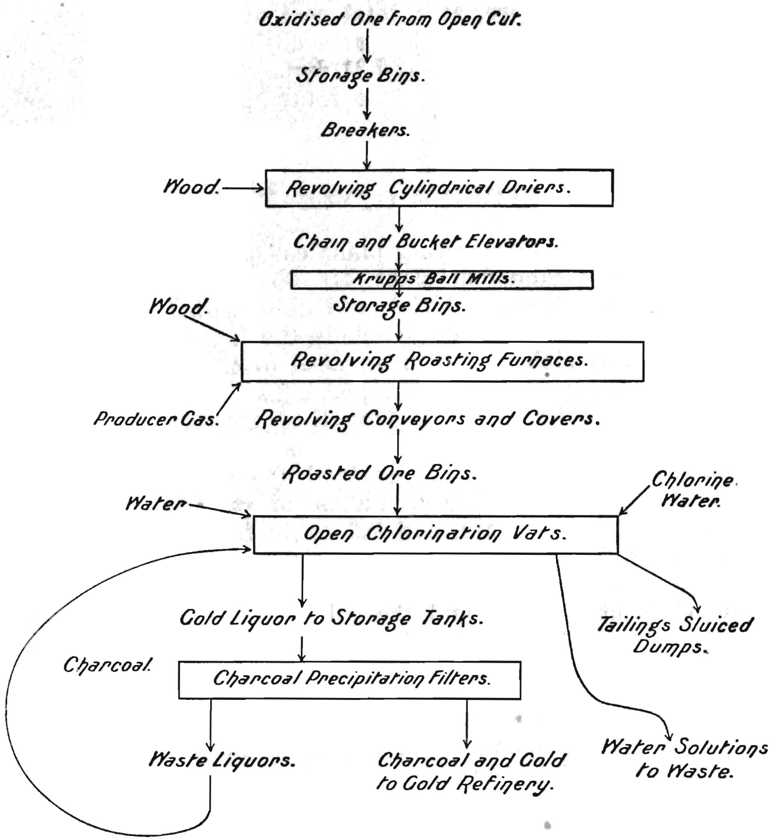
The gold-bearing solution is divided up, and each portion is run through one filter. No trouble is experienced with the precipitation of the gold. The solutions coming from all the filters unite and pass through two guard filters packed with charcoal like the main filters. These guard filters are double the size of the main filters, and afford an extra chance for any gold to be precipitated. The whole of the charcoal in the main filters is removed once a month and sent to the gold treatment plant, and the charcoal from the guard filters is shifted up into the main filters. the deficiency being made up with fresh charcoal. The sand on top prevents any dirt, etc., from contaminating the charcoal.

The average strength of gold solution entering the filters is 0.5 grain per gallon, the waste liquors show an average value of 0.0009 grain per gallon. Working costs per ton:—

	s.	d.
Crushing	3	5½
Roasting	3	7¾
Chlorinating	3	1½
Total	10	2¾

Total number of employees at time of my visit, when one section only was working, was sixty-seven.

— Flow Sheet —



The following statement shows the amount treated and the yield of gold from 1902 to end of 1909:—

Year Ending	Tons Treated.	Yield oz.	Average Value. dwts. per ton.
May 31st, 1902	117,021	61,973	10·59
,, 1903	112,487	70,456	12·52
,, 1904	119,833	46,970	7·83
,, 1905	134,435	56,604	8·42
,, 1906	135,210	52,884	7·82
,, 1907	120,373	35,313	5·86
,, 1908	123,469	21,010	3·40
,, 1909	108,191	17,236	3·19
,, 1910	52,917	12,287	4·64

Since my visit, the West Works have closed down. At the annual meeting of the Mt. Morgan G.M. Co., July 8th, 1910, the Chairman of Directors made the following remarks:—

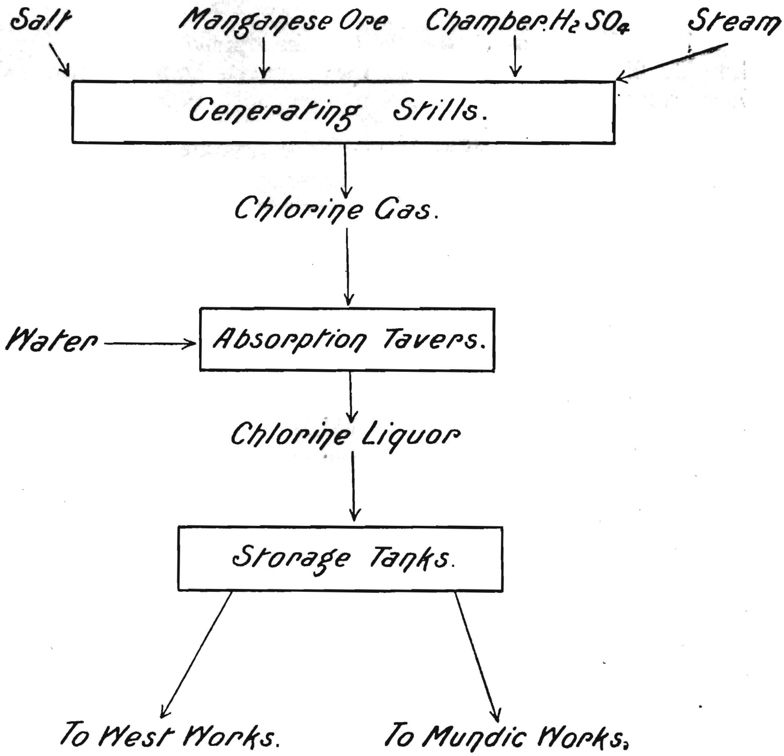
"The plant commenced operations in 1896, and the construction, with additions, cost £136,267, and it has treated 1,503,908 tons of ore, at a total cost of 15s. 5d. per ton, including cost of construction and all charges. The average grade of the ore treated was 7.31 dwts., and the average profit was 15s. 7½d. per ton, and the total profit from that source amounted to £1,175,422."

THE CHLORINE GENERATING PLANT.

The chlorine generating plant consists of three octagonal stills, built of sandstone slabs, 4ft. 6in. by 2ft. 9in., with top and bottom of same material, and lined with 6in. of concrete. Each still has a luted charge-hole in the top, and at the bottom of the front slab is a rubber-faced discharge door. Steam, under a pressure of 150lbs. per square inch, is led in on either side of the discharge door, through vulcanite pipes, and heats up the contents, and at the same time keeps the charge in agitation and obviates any tendency to pack or set. The charge per still consists of 120lbs. of salt, 118lbs.-136lbs. of manganese dioxide ore, and 2.6 cubic feet of sulphuric acid at 123deg. Twaddle. The salt and manganese ore are crushed fine, and then intimately mixed together. Then charged into the stills at the top, and the sulphuric acid run in through a trapped lead pipe. Half an hour after charging steam is admitted, and gradually turned on in increasing quantities until the charge is worked out, which usually takes about 1½ hours. The evolved chlorine leaves the stills at the top through a 4in. earthenware pipe into an earthenware main, and thence to scrubbing towers. These towers (two) are built of earthenware pipes, 2ft. inside diameter, encased in brickwork, and are 25ft. high. These towers are packed with stone for the first 10ft., and thence to the top with coke, over which fresh water is constantly trickling. The chlorine gas is led in at the bottom of one tower, and in ascending meets the descending stream of water and is partly absorbed. The balance is led away through a pipe from the top of this tower to the bottom of the second, where the remainder of the chlorine is absorbed. The chlorine liquor is led away from the foot of the towers to the storage tanks above the vats.

The manganese ore used is obtained from Mt. Miller, the salt from near Adelaide, S.A., and the sulphuric acid is made locally on the works.

FLOW SHEET



THE SULPHURIC ACID PLANT.

Chamber process used. There are seven chambers, total capacity of which is 100,000 cubic feet. Only six chambers are used, the other being under repair, etc. The chambers are 50ft. and 90ft. long, 20ft. wide, and 14ft. high. The floor and walls of the chambers are constructed of sheet lead. The sheets are connected by fusing the edges by means of an hydrogen flame. This method of joining lead is known as autogenous soldering. If solder is required pure lead only is used, as the presence of another metal gives rise to rapid corrosion of the lead, on account of galvanic action being set up. The floor of the chamber is simply a large leaden cistern,